MINING METHODS AND COSTS AT THE HAYDEN CREEK MINE OF ST. JOSEPH LEAD CO., ST. FRANCOIS COUNTY, MO.

BY MARVIN E. LANE

* * * * * * * * * **Information Circular 7781**

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INTRODUCTION

This paper, which describes the mining methods and practices at the Hayden Creek lead mine of St. Joseph Lead Co. in southeast Missouri, is one of a series being prepared by the Bureau of Mines on mining methods, practices, and costs in various mining districts of the United States.

The Hayden Creek mine is in the W 1/2 sec. 7, T. 36 N., R. 4 E., St. Francois County, Mo., about 1 mile southwest of the town of Frankclay (fig. 1). The mine is on the west edge of the mining district known as the Lead Belt. However, neither the ore body nor the underground workings are connected with those of any other mine in the district.

In contrast with most operations in the district, underground haulage is entirely trackless, and nearly complete delineation of the large, well-defined ore body by drilling permitted comprehensive planning of development and mining in advance of shaft sinking. The ore differs in character and occurrence from that of the main ore deposits in the Lead Belt in that the lead mineral, galena, occurs in the dolomitic matrix enclosing granite boulders instead of in dolomite limestone.

ACKNOWLEDGMENTS

Acknowledgment is made to the St. Joseph Lead Co. for its cooperation and for permission to publish the information in this paper. Special acknowledgment is due the mining, geological, engineering, and safety departments of the company's Southeast Missouri Division for their assistance in the preparation of data and illustrations.

HISTORY[3]

The lead deposits of southeast Missouri were known to the French as early as 1700. Explorers learned of their existence from Indians who had worked the surface deposits and transported quantities of lead to settlements along the Mississippi River.

A French expedition headed by Renault and LaMotte entered the region in 1717 to develop its lead mines. Small-scale mining was begun shortly thereafter at Mine LaMotte in Madison County. Lead mining had extended northward into Washington County by 1726, but operations ceased in 1731 when the French Crown canceled the charter of the bankrupt Mississippi Co. under which Renault had been working. Mining thereafter was carried on in a desultory manner until about 1799, when Moses Austin, a skilled

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[3] Winslow, Arthur, Lead and Zinc Deposits: Missouri Geol. Survey, vol. VI, 1894, pp. 267-302.
Cornell, Irwin, H., How the St. Joseph Lead Co. Grew: Min. and Met., vol. 28, No. 488, August 1947, pp. 362-365.

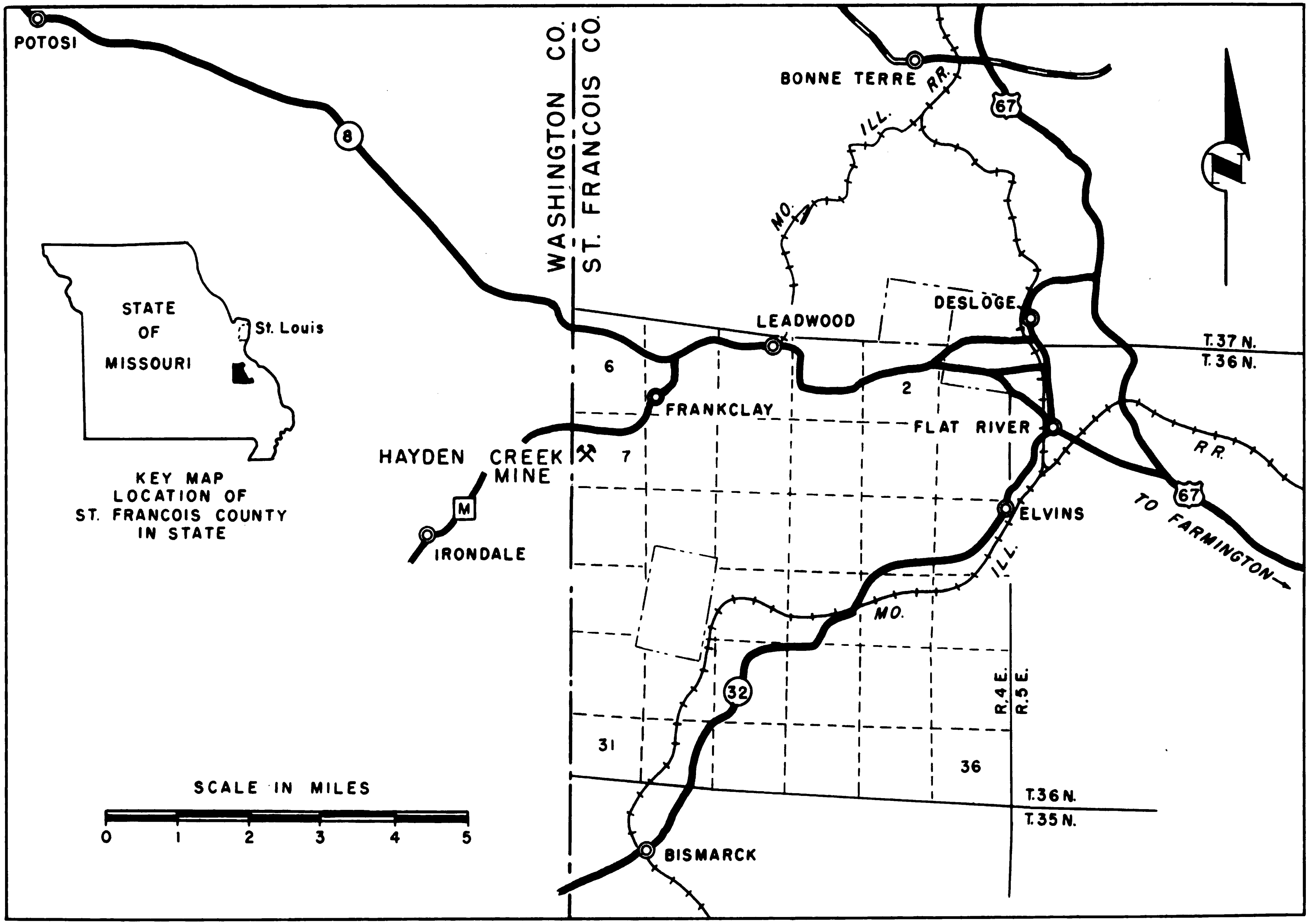

Figure 1. - Location map—Hayden Creek mine, St. Francois County, Mo.

operator from Virginia, erected a shot tower and a sheet-lead-manufacturing works. Within a few years he increased the recovery of lead from the ores by building and operating a reverberatory furnace.

Lead output from the surface "diggings," about 1,000 tons a year between 1816 and 1818, was shipped to the riverside town of Herculaneum, where 3 shot towers had been built. Production had increased to about 1,500 tons of lead a year by 1825 and continued to increase until the beginning of the Civil War, when hostilities disrupted operation of the furnaces.

St. Joseph Lead Co. was organized May 25, 1864. Its name may have been derived from "Mine a Joe," an early-day mine near the present town of Flat River. The holdings of the newly formed company consisted of 946 acres of mining lands called the LaGrave Mines. The company began mining near Bonne Terre where rich lead ore was discovered underground and in 1869 produced 605 lead pigs (about 70 pounds each) monthly with hand jigs and charcoal hearth smelting. Output increased to 2,500 pigs a month by 1873.

Several other companies, including the Desloge Lead Co., Doe Run Lead Co., St. Louis Smelting and Refining Co., and the Federal Lead Co., were active in the Lead Belt for many years and contributed to the rapid expansion of mining in the district from the late 1800's to the early 1900's. The holdings of these companies were acquired over a period of years by St. Joseph Lead Co.

More than 9 million short tons of pig lead has been produced from ores mined in the Lead Belt. Except for the Broken Hill mines in Australia, southeast Missouri has yielded more lead than any other district in the world, and has supplied about a third of the United States output.

Peak production of metal was reached in 1925, when 208,915 short tons of pig lead was recovered from ores that ranged from 3.5 to 4.0 percent lead content - considerably richer than the ores mined in recent years. Peak mine output of crude ore was nearly 7 million tons in 1942. Output in 1955 was about 6 million tons.

The Hayden Creek mine is one of the newer operations in the Lead Belt. It is also referred to as the No. 22 mine because the shaft was the 22d to be sunk in the district by St. Joseph Lead Co. The ore body was discovered by random diamond drilling in 1943, but systematic drilling to outline the ore did not begin until 1948. Shaft sinking began in May 1949 and underground development in July 1951.

DESCRIPTION OF DEPOSIT[4/]

The character and occurrence of the Hayden Creek ore body are unique in the Lead Belt. The ore occurs in a cemented granite boulder bed on the flank of a Precambrian granite knob (fig. 2), whereas most of the ore mined in the Lead Belt occurs in disseminated and bedding-plane deposits in the Bonneterre dolomite.

The boulder bed consists of two units separated in part by a tongue of sandstone. The lower unit, which has a maximum thickness of 15 feet, is a basal conglomerate that lies directly upon the granite surface. Overlying this unit is the

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[4/] Ohle, E. L., Geology of the Hayden Creek Lead Mine, Southeast Missouri: Trans. AIMME, vol. 193, 1952, pp. 447-483.

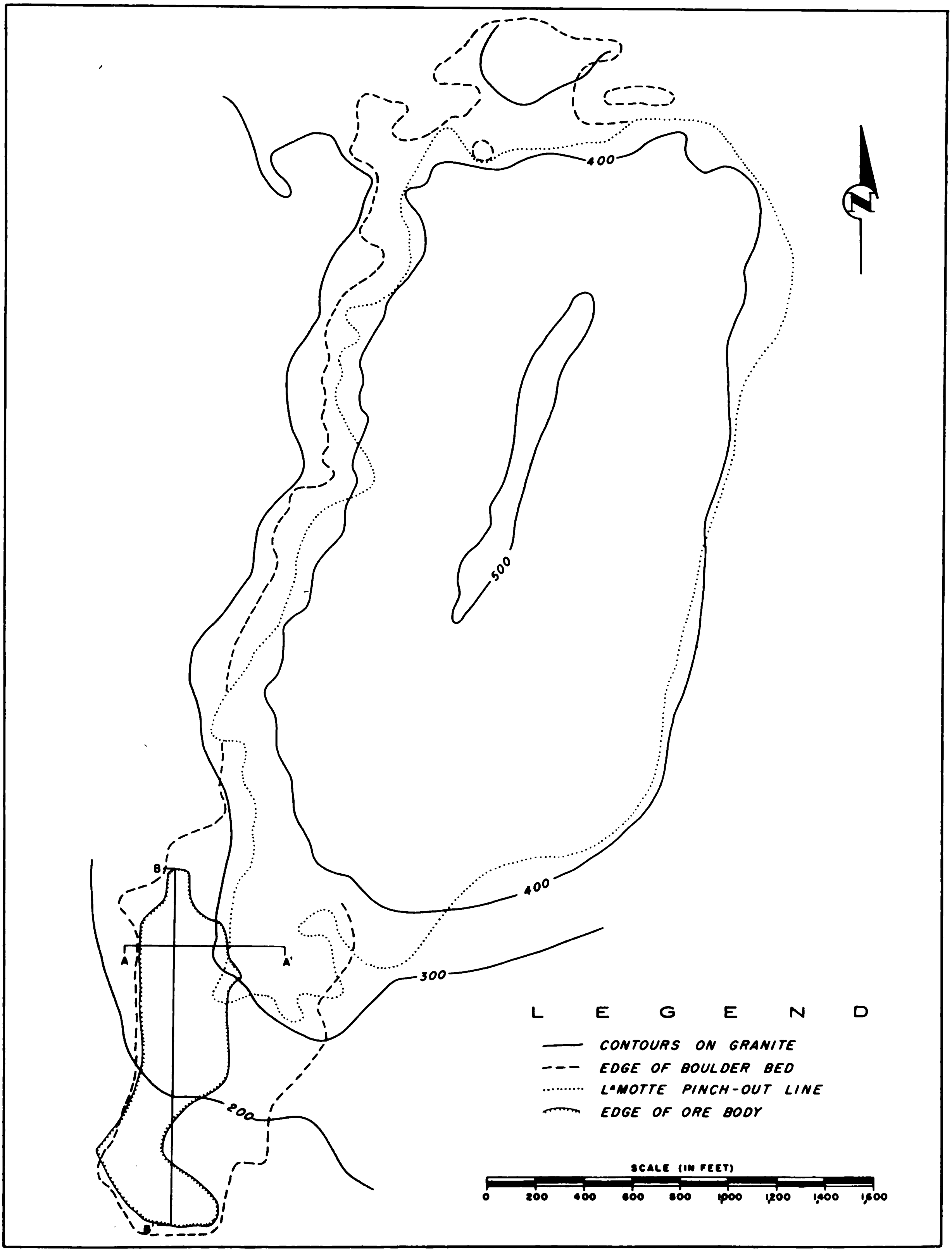

Figure 2. - Geologic plan of granite knob and boulder bed.

Upper Cambrian Lamotte sandstone, which pinches out along the flanks of the granite knob. The upper, main boulder unit extends out over the sandstone for many hundreds of feet south of the knob. This boulder mass, which has a maximum thickness of 115 feet, contains most of the ore (fig. 3). The Bonneterre dolomite overlies and in some places interfingers with the main boulder unit on the flank of the knob and overlies the sandstone beyond the outer margin of the boulder bed.

The boulder bed or conglomerate consists of unsorted granite boulders and fragments cemented by sandy dolomite (fig. 4). The large boulders range from 6 inches to several feet in diameter and are well-rounded. The presence of several rudely stratified boulder zones and lenses of dolomite within the boulder bed and the interfingering of Bonneterre dolomite with it indicate that the main boulder unit is a shoreline phase of the lower part of the Bonneterre formation.

Galena, the only ore mineral, mainly occurs as disseminated grains replacing the cementing dolomite and to a minor extent as fracture filling in granite boulders. Not all of the boulder bed is mineralized; the largest continuous ore zone lies just above the top of the Lamotte sandstone in the lower part of the main boulder unit and west of the sandstone "pinchout" (sec. A-A', fig. 3). Other smaller, less continuous ore zones occur higher in the boulder bed, and some drill holes showed considerable thicknesses of low-grade material between these upper ore zones and the main ore body. A small quantity of ore has been found in the basal conglomerate near the sandstone pinchout.

The Hayden Creek ore deposit is about 1,600 feet long and 400 feet wide and ranges in thickness from 9 feet to 80. The long axis is north-south, and the floor of the main body of ore slopes upward to the north at an average gradient of 2.5 percent. In the west-central part of the mine a small but minable ore body occurs about 50 feet above the haulage level.

METHODS OF PROSPECTING AND EXPLORATION

Prospecting, as defined by St. Joseph Lead Co., consists of geologic exploration and surveying, and test drilling. The customary first step in prospecting for lead deposits in virgin areas is to examine the surface geology for faulted areas and possible buried igneous highs.[5] The surface study is followed, where conditions appear to be favorable, by diamond drilling widely spaced holes to determine subsurface structures and whether or not the favorable host rocks have been mineralized. Trends of mineralization are explored and outlined by drilling at increasingly closer intervals.

The discovery of ore in the Hayden Creek area by prospect drilling was followed by systematic diamond drilling to determine the extent and nature of the ore. The deposit was drilled ultimately on 100-foot centers until the entire ore body was outlined. Core recovery was poor owing to the variability of alteration in the granite boulders and the difference in physical properties of the boulders and the dolomitic matrix. Mining has proved that the ore is thicker than was indicated by diamond drilling.

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[5] Wagner, R. E., "Lead Belt" Geology: Min. and Met., vol. 28, No. 488, August 1947, pp. 366-368.

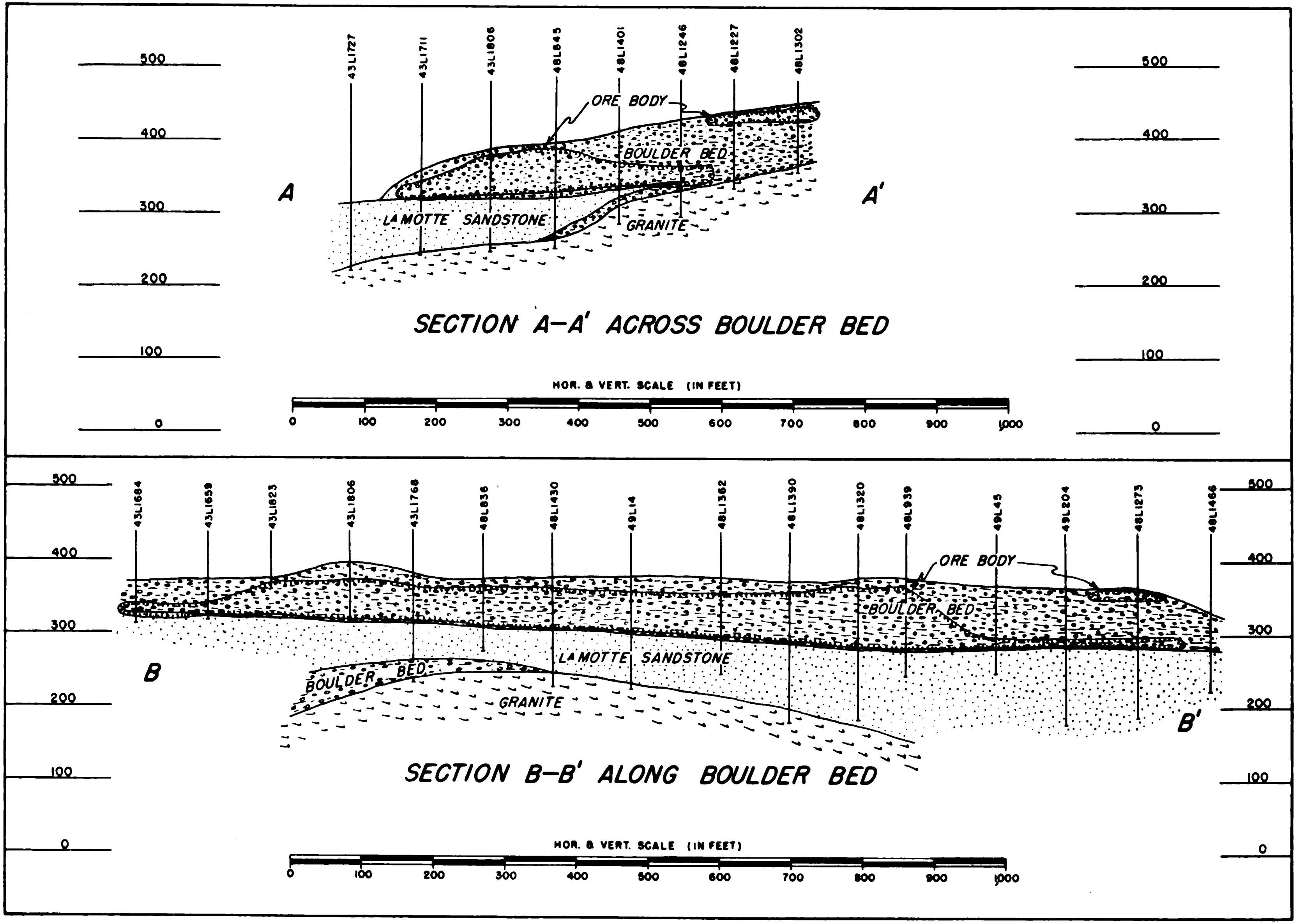

Figure 3. - Geologic sections through boulder bed.

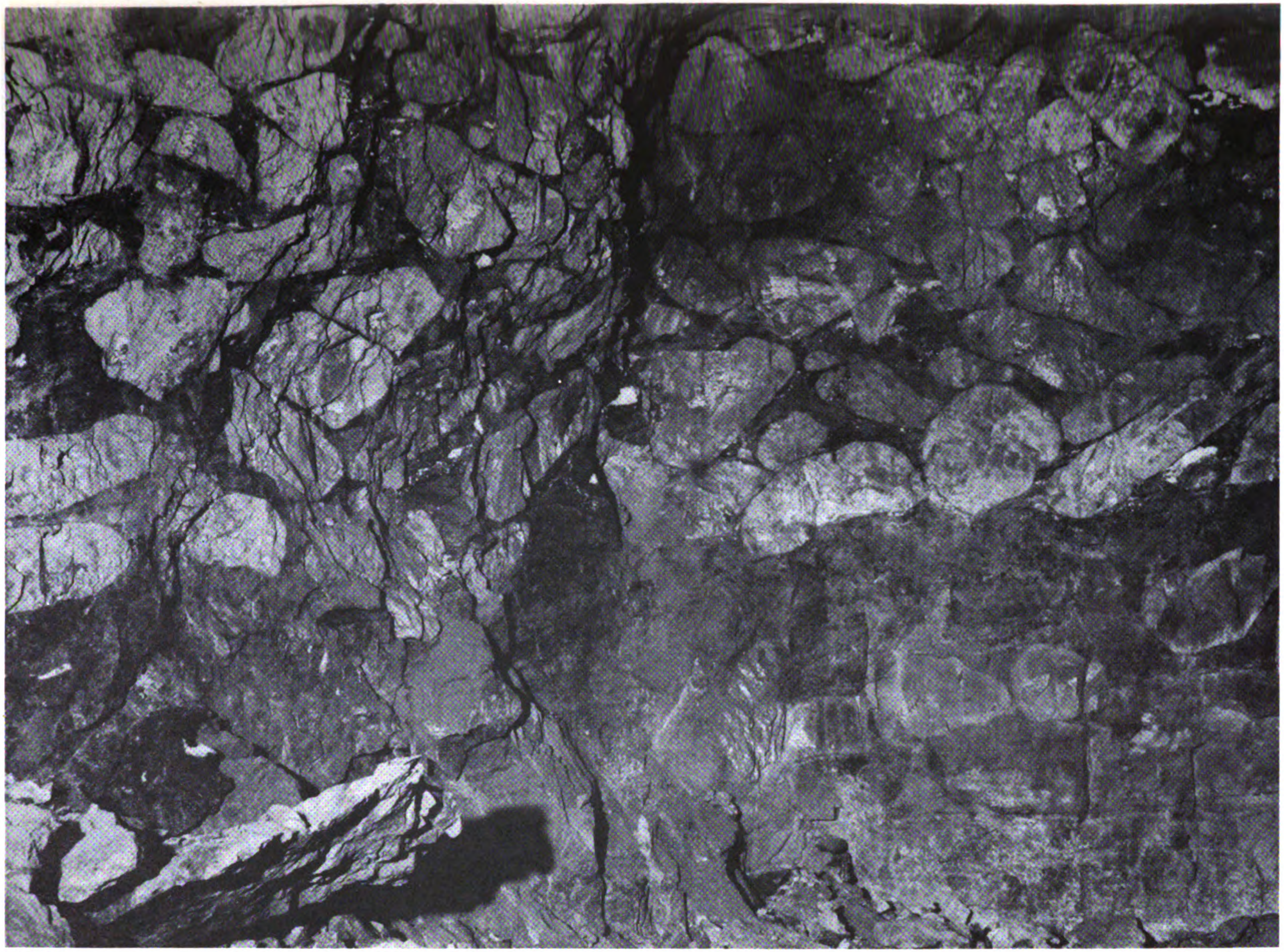

Figure 4. - Face of heading in boulder-bed ore. *(Courtesy St. Joseph Lead Co.)*

METHODS OF ESTIMATING GRADES AND TONNAGES OF ORE

The original estimate of ore reserves at Hayden Creek, completed January 1, 1949, from diamond-drill-hole data, indicated that the deposit contained 28,840 short tons of lead. The reserve estimate of October 1, 1955, plus the quantity of lead in ore mined to that date, showed an indicated total lead content of 28,670 short tons. The adjusted data are shown in table 1.

TABLE 1. - Production and ore estimate to October 1, 1955

	Ore, short tons	Grade, percent Pb	Lead in ore, short tons
Mined to date	828,496	2.09	17,314
Unmined (estimate)	507,000	2.24	11,360
Estimated totals	1,335,500	2.15	28,670

The grade and tonnage of ore mined are determined from analyses and weights of ore delivered to the mill. The daily average grade of ore received at the mill is prorated back to each of the active mine headings on the basis of the shift foremen's daily estimates of grades. Ore tonnages are prorated according to types of haulage equipment and number of loads hauled. The adjusted data are used to determine the quantities of ore and lead produced from the stopes or headings. At the end of each year the ore reserves are reevaluated by checking the remaining ore faces, deducting the quantity of ore produced, and adding the ore discovered during the year.

DEVELOPMENT

Access to the mine is through a vertical shaft sunk in the country rock 40 feet from the west edge of the ore body. The ore pocket and underground crusher room were cut in the rock between the shaft and the ore body. Development drifts, 9 by 12 feet in cross section, were driven into the ore from the shaft station. Only one haulage level was cut. The upper, smaller ore body was developed by driving two inclined raises from the haulage level to the ore. After a connection had been made between the raises on the upper mining level, one raise was used as an ore pass and the other for access of men, equipment, and supplies.

Shaft Sinking

The shaft is 9 by 18 feet in plan and has 3 compartments. The total depth from the top of the concrete collar to the bottom of the sump is 701 feet, and the depth to the mine floor is 613 feet (fig. 5).

Shaft sinking was begun in May 1949. The operation was directed by an outside contractor, but labor was supplied by the company. None of the crew had had previous shaft-sinking experience. Later, the knowledge and skill acquired in sinking the Hayden Creek mine shaft enabled the company to sink three shafts elsewhere in the district with a high degree of efficiency, at comparatively low cost, and with a near-perfect safety record.

Shaft sinking was on a 3-shift-per-day schedule. Each shift crew consisted of 8 men - 4 shaftmen, a hoistman, a topman, a dumpman, and a shift foreman. In addition, a maintenance man was employed on day shift, and a shaft foreman was in charge of the operation.

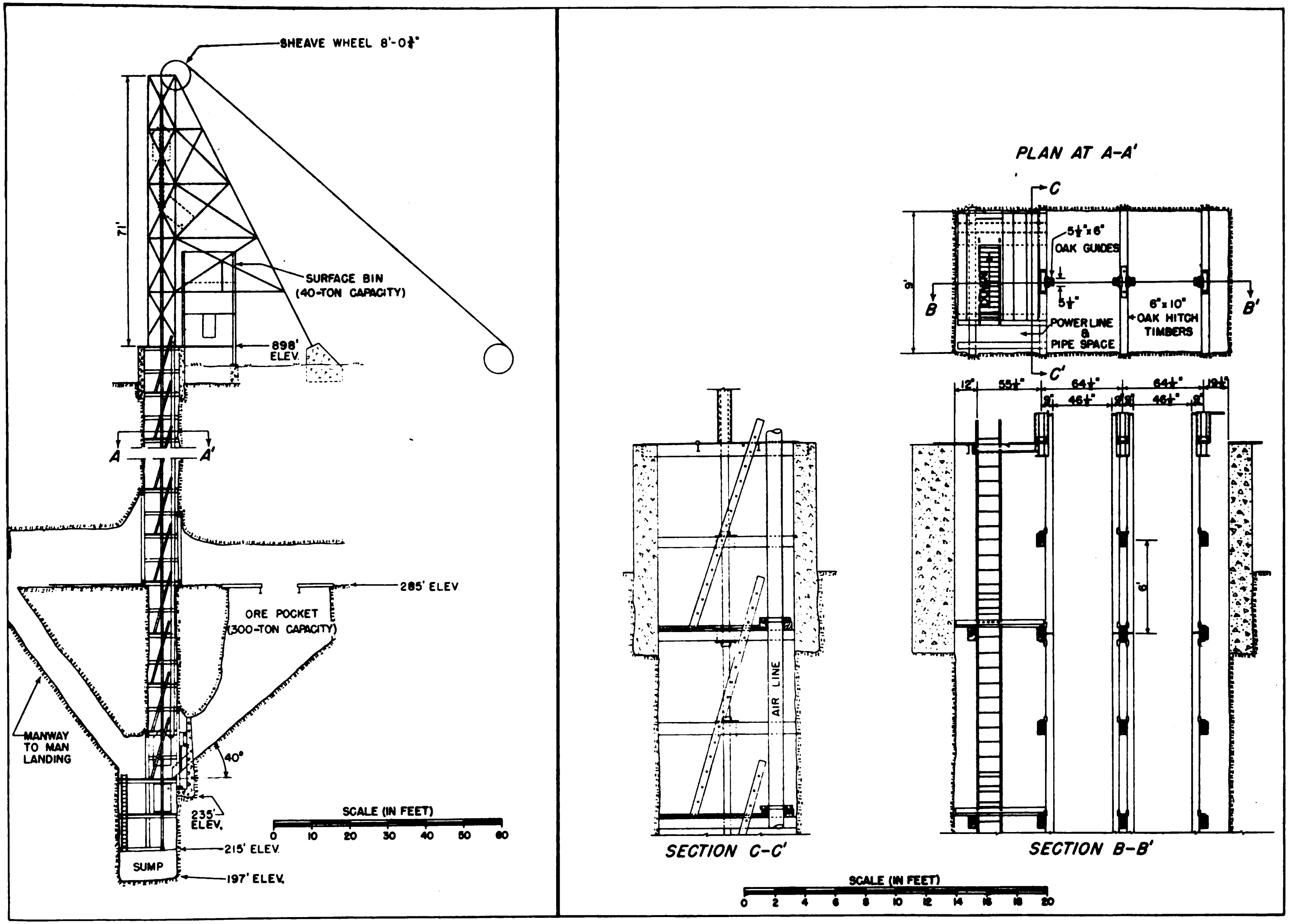

Figure 5. - Shaft and timbering layout.

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Surface installations and equipment included a temporary headframe, sinking hoist, and three portable compressors.

Standard center V-cut rounds were drilled to an average depth of 7 feet. Hand-held, 2-5/8-inch sinker drills, with 7/8-inch hexagon hollow carbon drill steel (1-inch by 4-1/2-inch shanks) and 1-1/2-inch detachable tungsten carbide crossbits, were used to drill the shaft rounds.

Semigelatin dynamite of 60-percent strength was used for blasting. Millisecond-delay electric blasting caps with 12-foot lead wires were connected in parallel multiple series and fired with 110-volt alternating current.

Mucking was done with a Riddell shaft-mucking machine, which loaded the broken rock into a sinking bucket for hoisting and disposal. The machine consists essentially of a clamshell bucket suspended from a movable carriage, supported by a steel frame. The carriage and bucket were operated by air motors.

The shaft was bottomed before timbering began. A section of the shaft in shale was gunited concurrently with the sinking, to prevent spalling.

Other than shaft sets installed at the haulage level and at the ore pocket, timbering mainly consisted of cutting timber hitches in the sides of the shaft at regular intervals; setting and securing the timbers (dividers) for support of skip guides, manway ladders, and pipelines; installing skip guides; and constructing ladder ways. The timber hitches were cut in a continuous operation from a double-deck drill platform suspended in the shaft. While one set of hitches was being cut from the lower deck, the next set was marked off from the upper deck. The hitches were cut on 6-foot vertical centers, except for those outside the manway compartment, which were 12 feet apart (fig. 5). Oak timber was used for shaft sets and guides. The shaft or "hitch," timbers are 6 by 10 inches and the guides are 5-1/2 by 6 inches.

The distribution of time required to complete the various phases of the shaft-sinking and auxiliary operations is shown in table 2.

TABLE 2. - Approximate distribution of time spent on shaftwork

Job	Number of days	Job	Number of days
Collar construction	25	Cutting hitches	25
Erecting temporary headframe ..	8	Installing timbers	35
Sinking	150	Cutting and installing pocket .	80
Guniting	36	Total	461
Drifting and raising	102		

Drifting

Underground development, which included excavation of a crusher room and the shop and supply rooms, was begun in July 1951 on a 2-shift schedule. Inasmuch as the nature and attitude of the ore body permitted mining with a minimum of preparation, development was directed primarily toward providing good drainage and ventilation for the mine.

In order to ensure adequate drainage from the start, a drift was driven to the lowest part of the ore body (southeast of the shaft), and a small sump was sunk there.

Another drift was driven northeastward from the shaft area to intersect a 24-inch churn-drill hole for ventilation. This drift was continued, bearing to the right within the ore body, to connect with the drift to the sump (fig. 6).

Drifting to develop the ore body for mining followed the pattern shown in figure 6. The initial development drifts were 9 feet high and 12 feet wide and were driven along the bottom of the ore body. The height was increased later to 10 feet to provide headroom for haulage trucks.

Much of the development work was done by 4 drift crews, 2 on each shift, under the supervision of a mine captain and a trainee shift foreman. Development was reduced to one crew on each shift in March 1953; suspended in June 1953; resumed in November by one crew per shift; and completed in May 1954. Subsequent development has been limited to driving a haulageway toward another ore body 3,000 feet to the northwest.

A normal cycle of operations was followed in driving the drifts. A 2-man crew first mucked out and removed the rock or ore broken at the end of the preceding shift, drilled out the next round, then loaded and blasted the round at the end of the shift.

Drift rounds were drilled by two 2-5/8-inch rock drills, with 42-inch air feeds, mounted on 2-inch columns. The drill steel and bits were of the same type and size as those used in shaft sinking. Drill steel was made up in lengths suitable for 3-foot changes.

The standard drill round used for advancing the 9- by 12-foot drifts was a modified pyramid-cut round of 30 holes (fig. 7). Depths of the rounds ranged from 5 to 8 feet, depending on the type of ground and time available for drilling. The holes were loaded with 45-percent semigelatin dynamite. Regular-delay blasting caps with 8-foot lead wires were connected in a single series, and the charge was detonated by a blasting machine. Powder consumption was about 14 pounds per foot of advance.

Two-drum scraper hoists, powered by 50-horsepower, 440-volt motors, were used during the early stages of development to clean the drifts. The 54-inch scrapers, pulled by 5/8-inch, 6 by 19 steel wire ropes, scraped the broken rock to the ore pocket. As soon as the development openings had been advanced far enough to permit the use of mobile trackless equipment, Joy loaders and 12-1/2-ton electric shuttle cars replaced the scrapers. The shuttle cars were replaced in 1953 by 7-1/2-ton diesel-powered Tournarocker trucks.

Raising

Raises to the upper-level ore body were inclined at angles ranging from 40° to 55°. They were driven in the same manner as the drifts, with regular drift crews and drilling equipment.

Development Slabbing

Side or back slabbing, as applied to development, was done mainly to enlarge a drift or other development opening to the extent that it could be used as a haulageway or for installation of stationary equipment. Bottom slabbing was done to maintain haulage grades and to build, widen, or deepen drainage ditches.

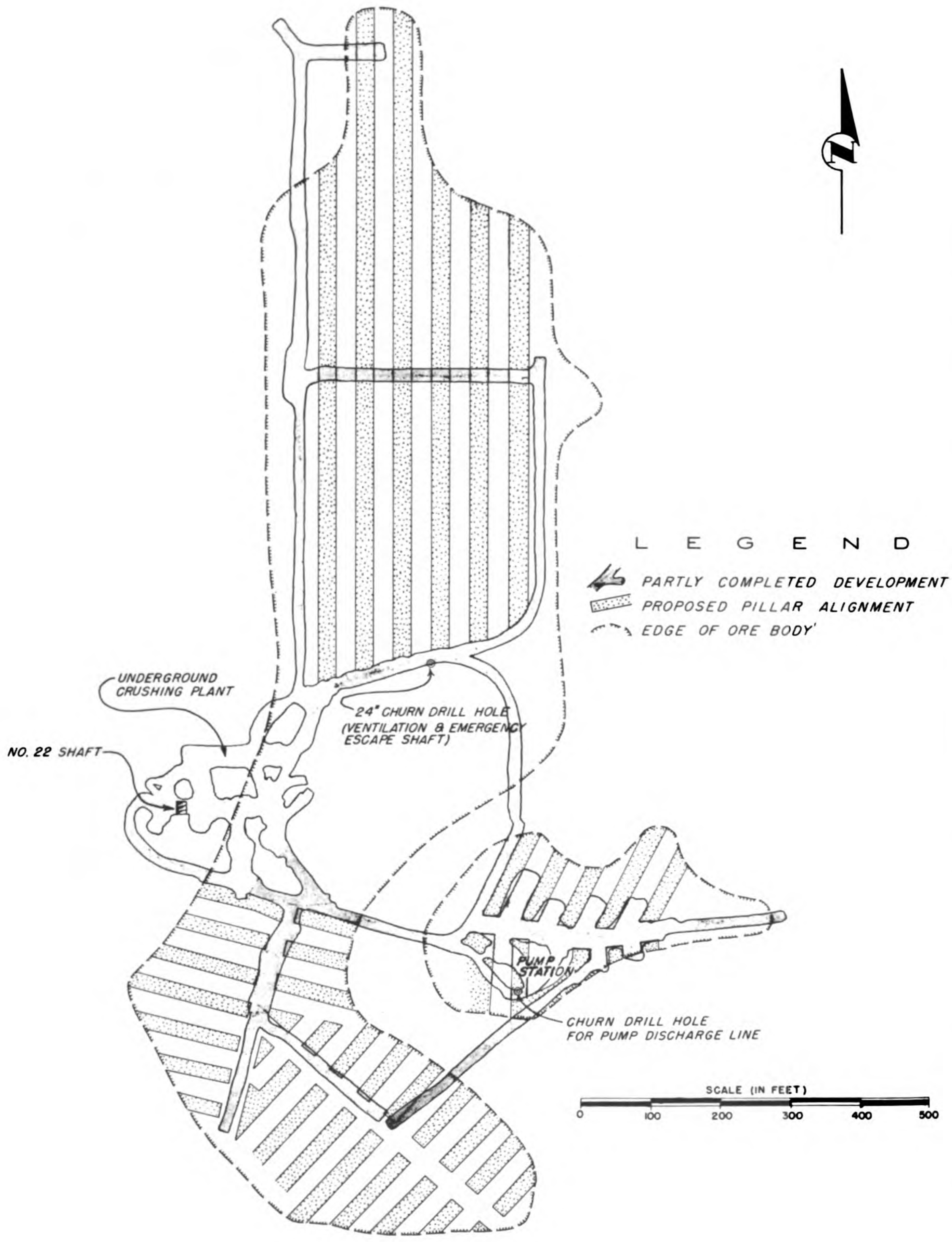
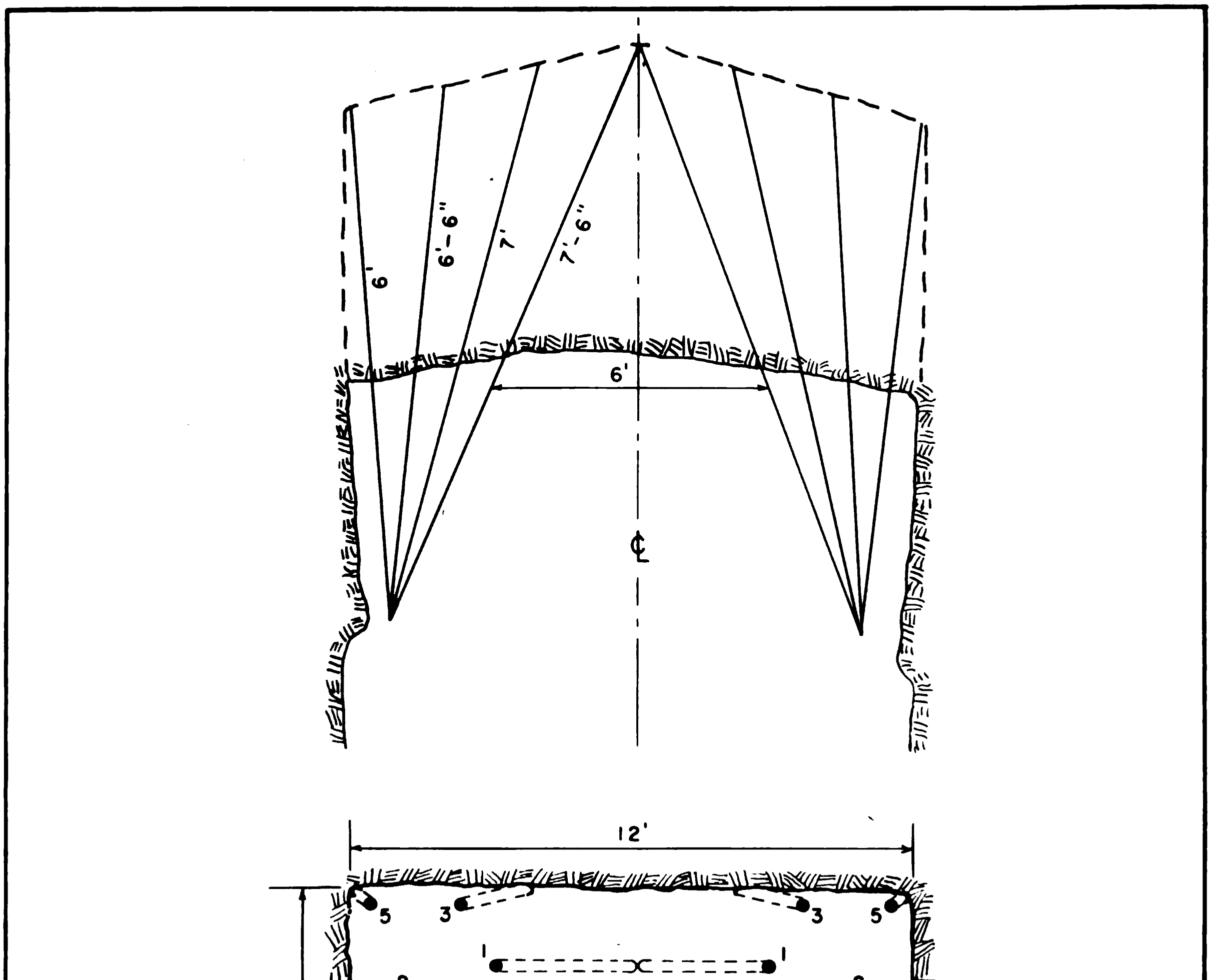

N
LEGEND
PARTLY COMPLETED DEVELOPMENT
PROPOSED PILLAR ALIGNMENT
EDGE OF ORE BODY
UNDERGROUND CRUSHING PLANT
NO. 22 SHAFT
24" CHURN DRILL HOLE (VENTILATION & EMERGENCY ESCAPE SHAFT)
PUMP STATION
CHURN DRILL HOLE FOR PUMP DISCHARGE LINE
SCALE (IN FEET)
0
100
200
300
400
500

Figure 6. - Layout of proposed plan of mining, showing partly completed development.



NUMBERS DENOTE ORDER OF FIRING OF DELAYS
("0" DENOTES INSTANT)

Figure 7. - Standard 9- x 12-foot drift round (pyramid cut).

For record and accounting purposes, all slabbing that is done in connection with development work in the Lead Belt is expressed in terms of "equivalent drift footage." An equivalent drift foot is defined as the amount of slabbing accomplished with the same effort as that required to drive 1 foot of drift. Slabbing is done with the same equipment as that used in drifting.

MINING

Mining was begun on a small scale in July 1952. The rate of output had increased to 1,000 tons of ore per day by June 1953 and averaged 1,303 tons per day during the last 6 months of the year. Production was cut back in October 1954 to a daily average of 1,000 tons and has been maintained subsequently at that rate. All mining has been done on a 2-shift schedule.

The room-and-pillar method was adopted for mining the Hayden Creek ore body. Mining progressed according to the layout in figure 6 and the plan of workings in figure 8. The rooms, which are 26 feet wide between 20-foot pillars, were driven along lines set by the surveying crew.

Rooms were advanced upgrade along the floor of the ore body, and pillars were turned by crosscutting at specified intervals. Mining was limited at first to a height of 9 feet, regardless of the thickness of ore. This restriction was imposed because: (1) The reaction of the boulder conglomerate to mining was unknown at the time, and limiting the height of openings to 9 feet permitted close checking of the back and side walls; and (2) cost data for the V-cut stope round method of breaking ore were desired for comparison with costs for the side-slabbing method that is commonly used in the Lead Belt. The restriction of room height to 9 feet was continued until February 1953 when the limit was raised to 10 feet. This limitation was removed later in the year, and headings were driven thereafter up to 14 feet in height.

All drilling, during the period when mining was confined to advancing the room headings, was done with mobile, rubber-tired drill jumbos. The jumbos are operated by air motors and are equipped with two hydraulic boom arms, each of which carries a 2-7/8-inch rock drill mounted on an 8-foot chain feed (fig. 9). Drill steel and bits are of the same types as those used in development.

A standard V-cut jumbo stope round of 36 holes (fig. 10) was drilled for breaking ore in the headings. The rounds were pattern-drilled, and the various positions of the boom arms were marked to eliminate guesswork in directing the holes. Drill steel of 2 lengths - 7 and 14 feet - was used to drill the rounds. Ore breaking with the jumbo round averaged 76.7 tons per drill shift in the second half of 1953.

"Back slabbing," a term used in the Lead Belt to denote the method of mining ore from the backs of stopes or rooms, began in January 1954. Nearly 44 percent of the ore produced from then to October 1, 1955, was mined by this method. At Hayden Creek drill jumbos and column-mounted drills are used for back slabbing. With both types of equipment, rounds of upholes are drilled first and blasted; then the machines are set on top of the broken ore, and slabbing continues to the limits of the ore. Figure 11 is a section across the mine workings and ore body where part of the ore has been mined by back slabbing.

Semigelatin dynamite of 45-percent strength is used for ore breaking. Millisecond-delay blasting caps with 12-foot lead wires are connected in a single series and fired by a blasting machine. The holes are loaded and blasted by the driller as part of the work of his shift.

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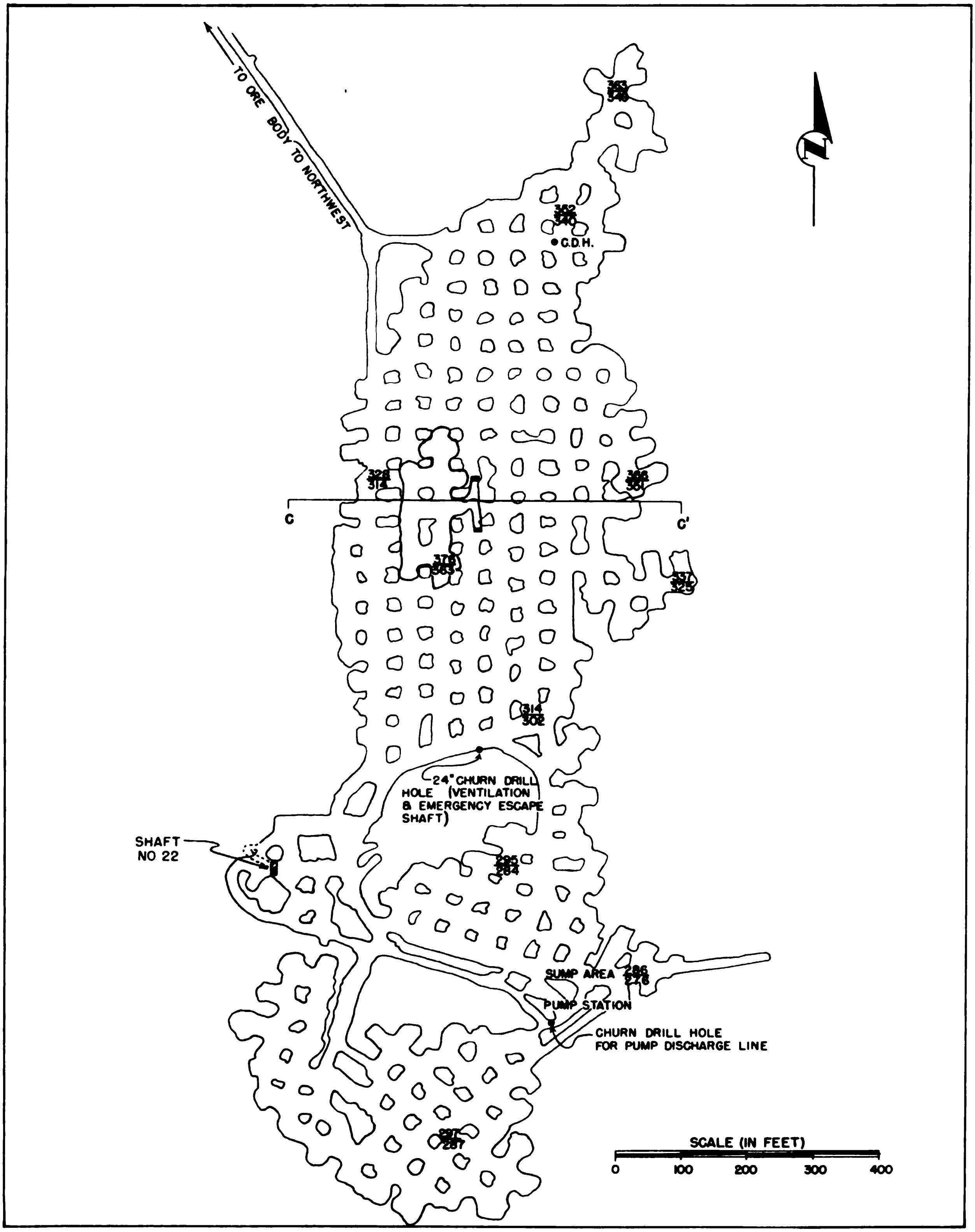

Figure 8. - Plan of mine workings.

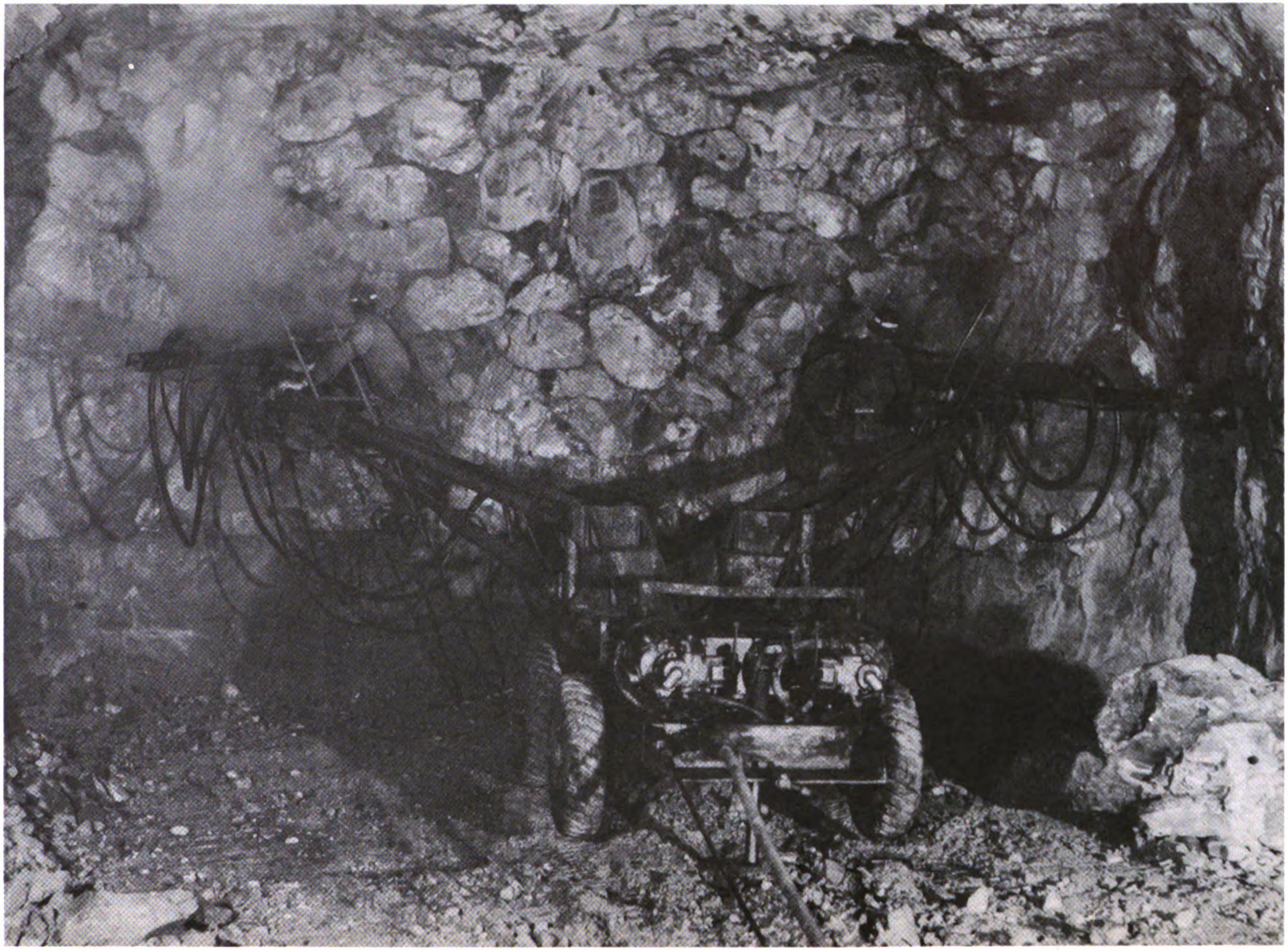

Figure 9. - Two-drill jumbo. *(Courtesy St. Joseph Lead Co.)*

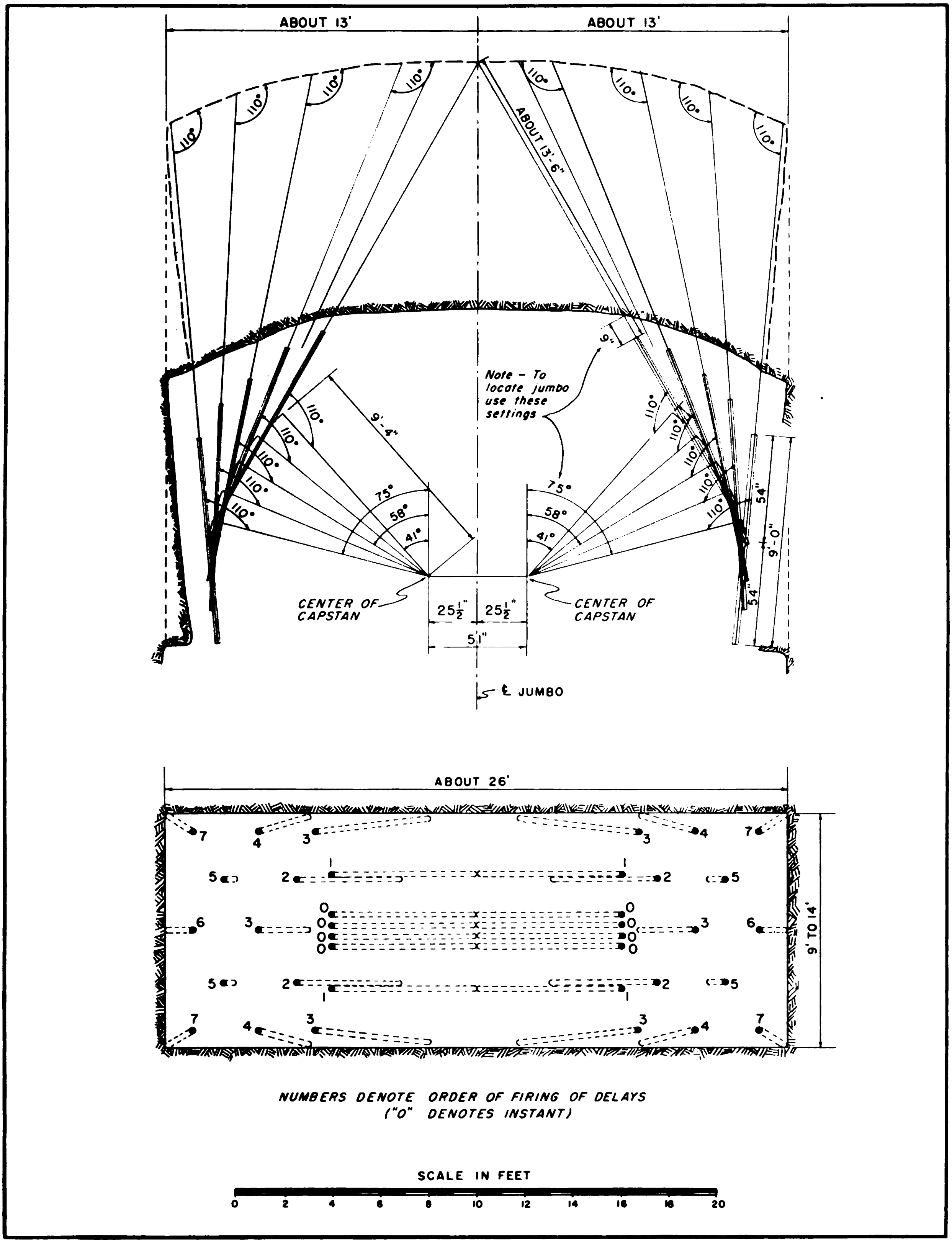

Figure 10. - Standard V-cut jumbo stope round.

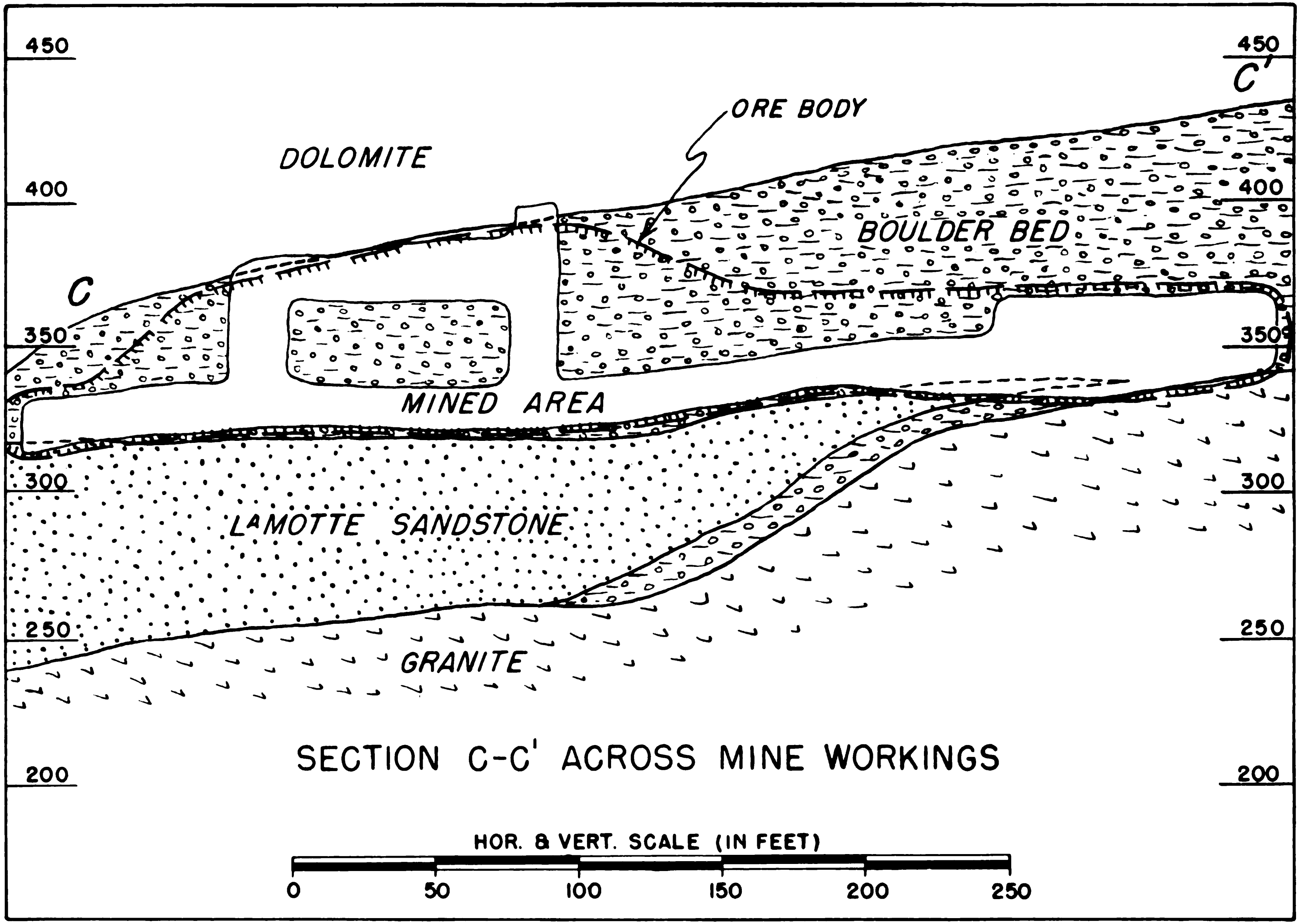

Figure 11. - Section across mine workings (see fig. 8).

No support other than the pillars has been required for stopes within the boulder conglomerate. The backs and walls of the haulageways and active headings are checked periodically, and all loose rock, regardless of size, is barred down. Most of the loose material has been small. In drifts and rooms where the back is dolomite, unstable roofs are supported by 4-inch channel irons pinned by 1-inch roof bolts. Lengths of bolts vary according to thickness of the unsound strata, but 10 feet is the length most commonly used. This method of roof bolting is in general use throughout the Lead Belt.

LOADING

All the ore produced in the Hayden Creek mine is loaded for underground transport with electrically operated Joy loaders (fig. 12). Alternating current is transmitted at 440 volts to the headings, where transformers convert it to 220 volts for operation of the loaders. Loading performance averaged 237 tons of ore per machine shift in the second half of 1953 and 286 tons in the first 9 months of 1955.

The ore mined on the upper level is moved by scraper hoists to the ore pass, where it falls to the haulage level. The scrapers are the same as those used in the early stages of development.

The ore is very abrasive so the wearing surfaces of the loaders are hard surfaced wherever practicable. A spare loader is kept in reserve for use when one of the operating machines is idled for general repair.

The weighted average unit costs of loading ore in 1953 were as follows:

Unit	Cost per ton
Operating labor	$0.080
Power	.005
Maintenance	.320
Total	.405

UNDERGROUND TRANSPORTATION

Rock and ore are transported underground entirely by trackless equipment. Electric shuttle cars (fig. 12) were used exclusively at first but were supplanted partly by diesel-powered Tournarocker trucks (fig. 13) in 1953. Since then the major part of the ore moved from the stopes to the underground crushing plant has been hauled by diesel trucks.

Although the rated capacity of the shuttle cars is 12-1/2 tons, the average load hauled at Hayden Creek is 10.8 tons. The diesel trucks have a smaller capacity, averaging 7.2 tons per load, but they are well adapted to haulage from narrow stopes where a short turning radius is required.

Each type of ore carrier has advantages over the other that make it more suitable for use under certain haulage conditions. The advantages of the shuttle car over the truck are (1) larger load capacity and (2) less headroom required. The diesel truck has the advantages of (1) faster travel - up to 15 miles per hour compared with 3 to 4 miles per hour for the shuttle car and (2) no limitation on haulage distance. The length of electric cable limits the shuttle car to a 1,400-foot haul; beyond that a double trolley line is required.

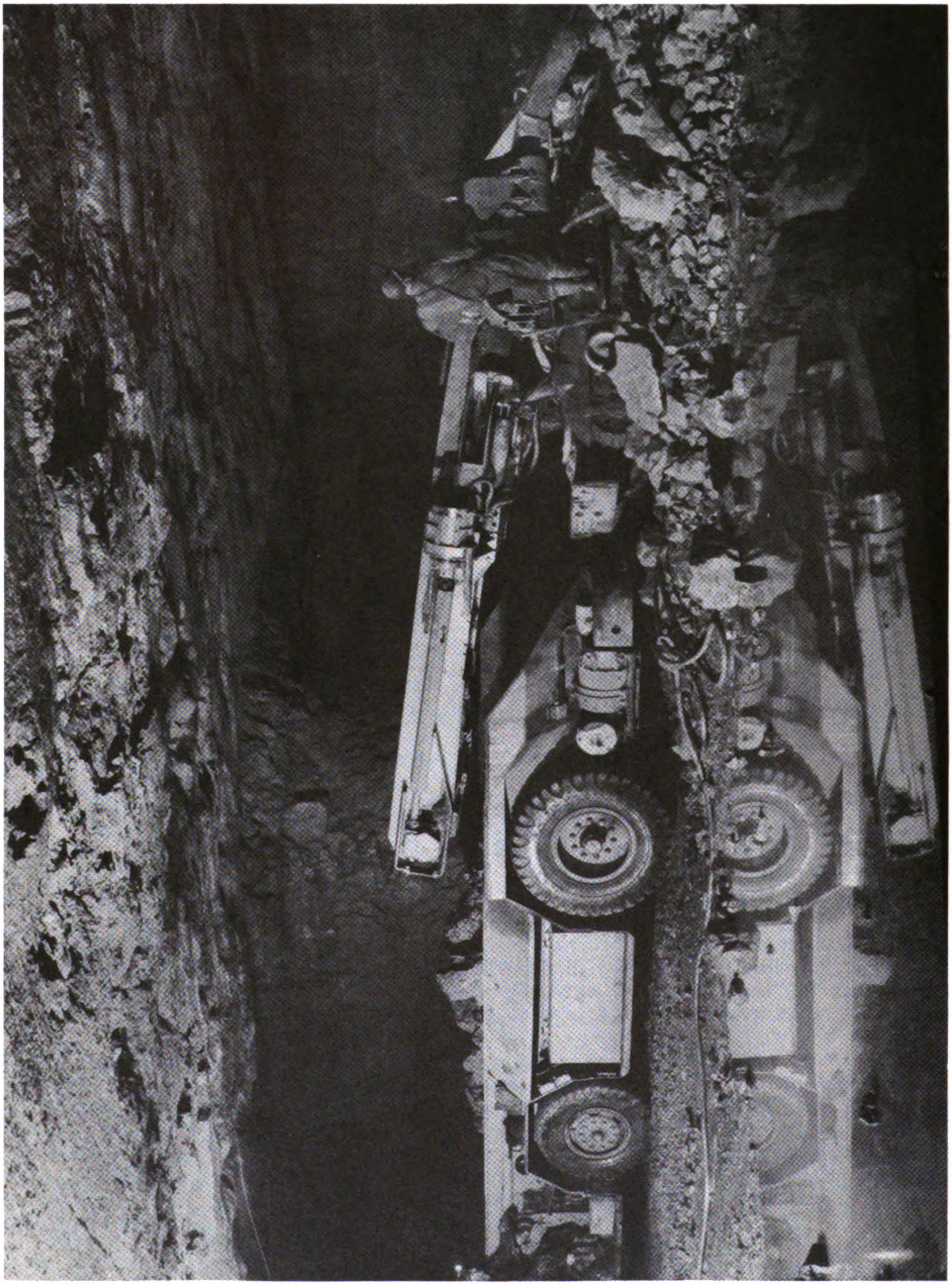

Figure 12. - Joy loader and shuttle car. *(Courtesy St. Joseph Lead Co.)*

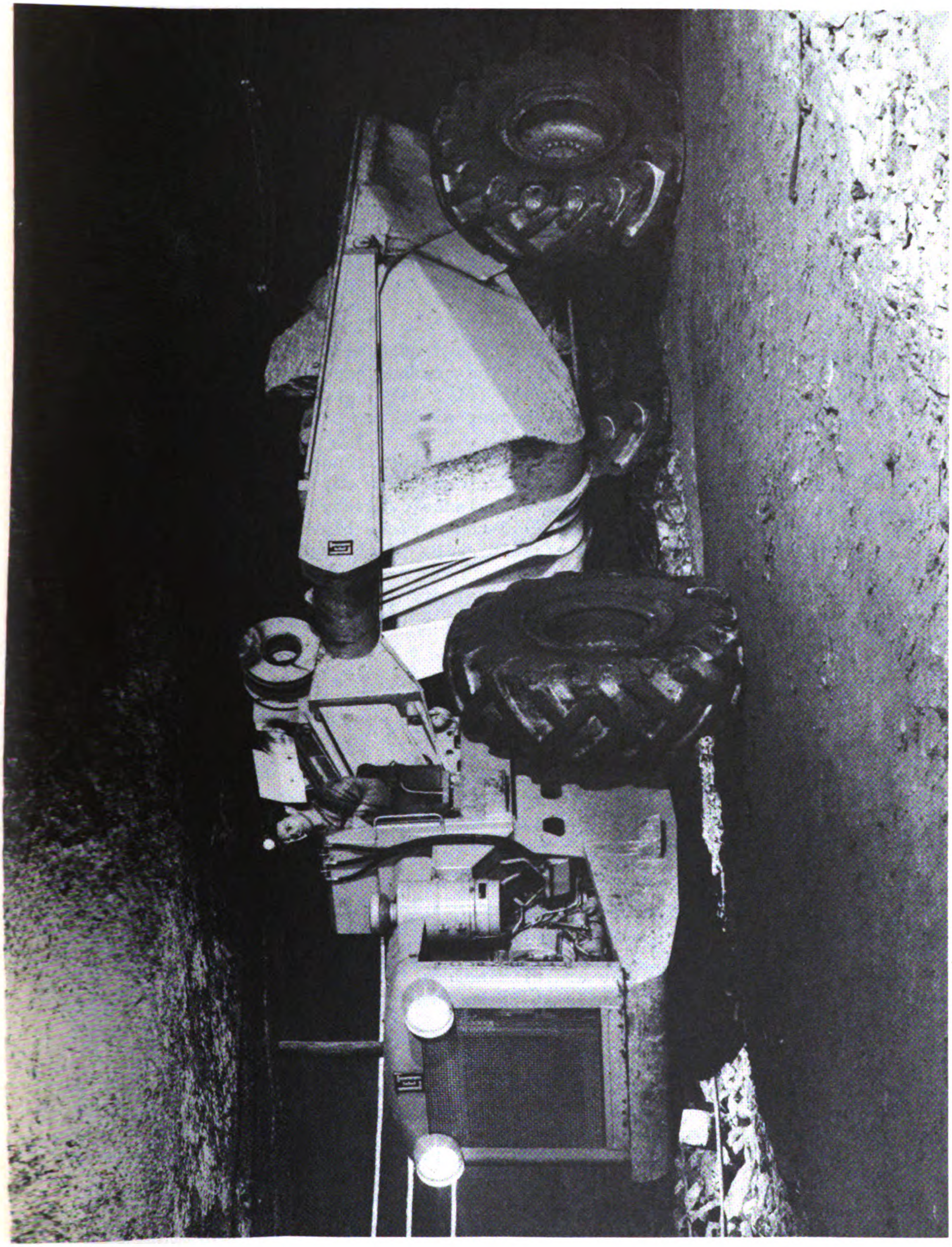

Figure 13. - Diesel truck. *(Courtesy St. Joseph Lead Co.)*

The average unit costs of transporting ore underground in 1953 were as follows:

Unit	Cost per ton
Operating labor	$0.080
Power, supplies and maintenance ..	.150
Underground roads	.010
Total	.240

A rubber-tired diesel tractor transports supplies underground and is used also for odd jobs.

Underground haulage roads are constructed of crushed stone and are graded periodically by a diesel tractordozer. Well-maintained drainage ditches keep the roadbeds dry.

Ore is dumped from the shuttle cars and trucks onto a 30-inch chain drag conveyer and is elevated thereby to a crusher. The crushed ore drops to a 30-inch belt conveyer and is moved to the skip pocket (fig. 14). The chain drag and belt conveyer are driven by a 40- and a 10-horsepower electric motor, respectively.

The skip pocket has a storage capacity of 300 tons. Undercut gates, operated by 12-inch air cylinders, control the loading of ore from the pocket into 4-ton, top-dumping skips. The skips are hoisted in balance, whereby 1 skip is being loaded while the other is dumping into the 40-ton surface bin. A cage for carrying men and supplies is mounted on the top of each skip. Each cage holds nine men.

The hoist is a 67- to 77-inch double conical drum, driven by a 250-horsepower electric motor. Hoisting cables are 1-1/8-inch, 6 by 19 steel wire ropes, and the hoisting speed is 766 feet per minute.

AUXILIARY OPERATIONS

Underground Crusher

Primary crushing is done underground by a New Holland 4040, double-impeller, impact breaker. The impellers are V-belt-driven by 75- and 100-horsepower, 440-volt electric motors. The crushing capacity is 150 tons per hour with a high ratio of reduction. The maximum feed size is 40 inches, and the size of the crushed product is minus-2-1/2-inch.

The advantages of this type of crusher and its installation underground are: (1) The maximum feed size of 40 inches makes it possible to handle many boulders that otherwise would have to be broken in the headings; (2) hoisting delays because of ore hanging up in the skip pocket are eliminated; and (3) the comparatively uniform size of the crushed rock minimizes spillage when skips are being loaded.

Underground Shop

An underground shop near the shaft is equipped for maintenance and overhaul of all operating equipment. An adjacent supply room stocks all replacement parts and materials. Overhauls and maintenance work are limited to replacement of parts except in emergencies. Replaced worn and reparable parts are reconditioned in other company shops.

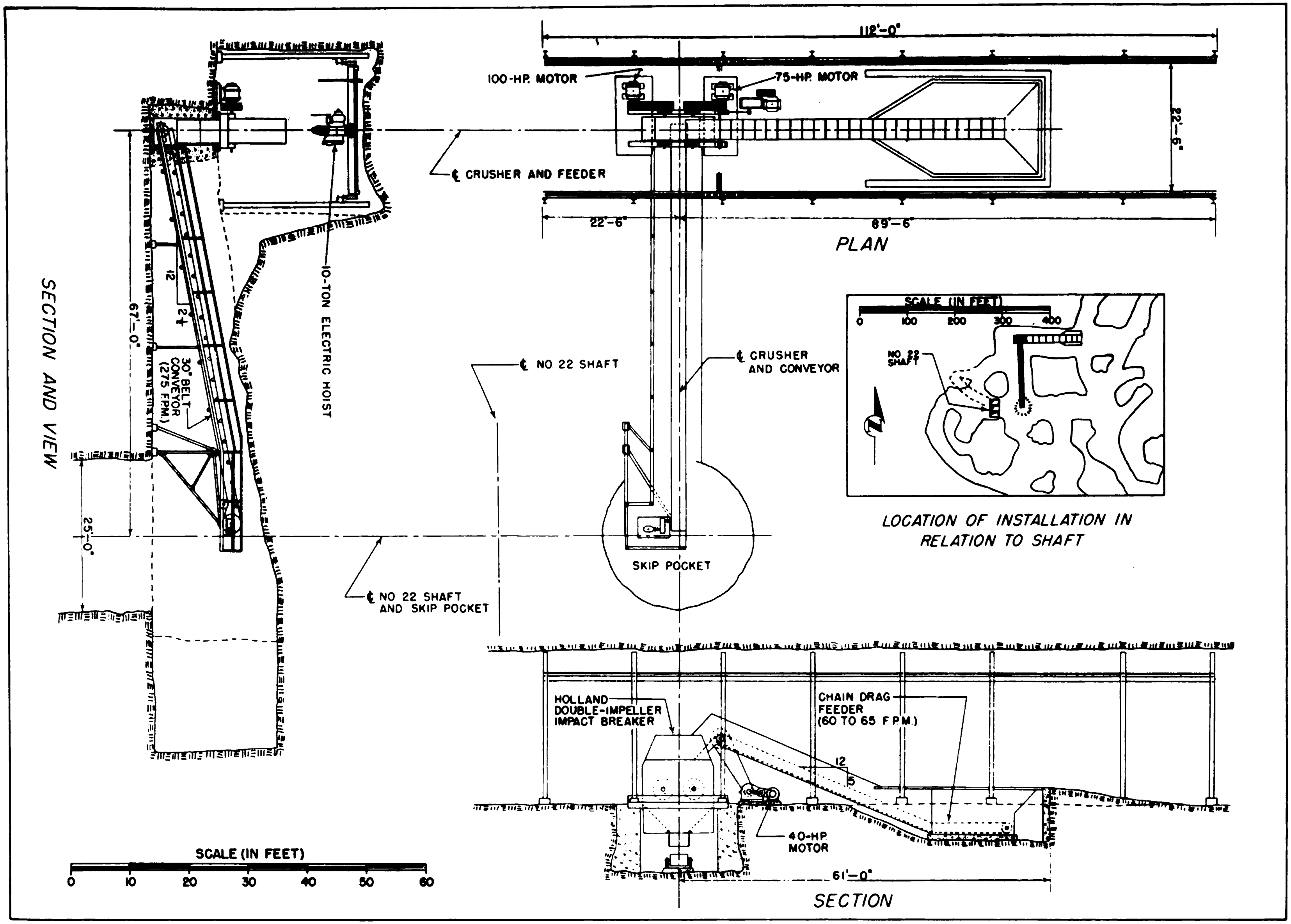

Figure 14. - Plan and sections of underground crushing plant.

Air Compressors

The compressor plant, which is installed in the surface hoistroom, consists of 1 operating 1,250-cubic-feet-per-minute air compressor driven by a 225-horsepower electric motor and a standby 1,000-cubic-feet-per-minute compressor with a 200-horsepower motor. Compressed air is produced at a pressure of 100 pounds per square inch and transmitted through the shaft into the mine in a 6-inch spiral-pipe air line. Four-inch lines branch off from the main line at intervals and reduce to 2 inches in the headings. The compressed air reaches the drills at a pressure of about 80 pounds per square inch.

Drill-Water Supply

Water for drilling is supplied by a 50-horsepower centrifugal pump at the pump station. The water is pumped through the mine in a 2-inch trunkline and to the headings in 3/4-inch branch lines. Water pressure at the pump is maintained at 180 pounds per square inch, which insures ample pressure and supply to the drills.

Motor Generator Set

A 235-horsepower, 440-volt a. c. motor-generator set in the hoistroom is used to generate 270-volt direct current for operation of the shuttle cars.

Escape and Ventilation Shaft

A 24-inch churn-drill hole, drilled from the surface several hundred feet northeast of the shaft and connected underground with a development drift (figs. 6 and 8), serves as a combination escape and ventilation shaft. The ventilating fan mounted over the drill hole can be moved readily, after removal of one bolt and clamp, to clear the hole for hoisting men from the mine.

The hoisting equipment consists of a small headframe erected over the shaft, a bullet-shaped 1-man cage attached to a 3/4-inch, 6 by 19 steel wire rope, and a 25-horsepower single-drum electric hoist. The depth from the surface to the mine level at the drill hole is 577 feet.

EXTRACTION AND PRODUCTION RATES

Approximately 83 percent of the minable ore is being extracted by breast stoping and back slabbing. It is expected that 75 percent of the remaining ore (in pillars) will be recovered later, indicating an ultimate extraction of over 95 percent.

The total output of ore will be somewhat greater than had been estimated at first, due to production of some low-grade ore that was not included in the original estimate. The additional tonnage will result from extending the stoping height in several areas and not from additional lateral developments. The original outline of the ore body, as determined by drilling, has proved to be quite accurate, except for minor local deviations.

Separate facilities for disposing of waste rock were not considered necessary when the shaft layout was planned, because virtually all development was to be done in ore. The waste rock broken in a haulageway that is being driven northwest to the separate ore body must therefore be dumped with the mined ore for crushing and hoisting. Ore dilution from this source averages about 10 percent.

Statistical data for development and mining since shaft sinking was begun are shown in table 3.

TABLE 3. - Development and production data

Development

Year	Feet of drift	Feet of raise	Feet of shaft	Equivalent drift feet of slabbing	Total feet of development	Ore produced, tons	Average grade, percent Pb
1949	-	-	591	-	591	-	-
1950	254	122	89	601	1,066	2,415	2.03
1951	714	128		1,051	1,893	11,873	1.79
1952	2,964	-		1,060	4,024	37,842	2.04
1953	1,282	76		452	1,810	12,091	1.49
1954 (to June 1).	88	299		326	713	5,500	1.00
Totals	5,302	625	680	3,490	10,097	69,721	1.82

Mining

Year	Breast stoping, tons of ore	Average height, feet	Grade, percent Pb	Back slabbing, tons of ore	Average height, feet	Grade, percent Pb	Total tons of ore	Average grade, percent Pb
1952	16,921	9.7	1.84	-	-	-	16,921	1.84
1953	250,843	11.8	2.48	-	-	-	250,843	2.48
1954	178,652	13.3	2.06	131,863	10.7	1.90	310,515	1.99
1955 (to Oct. 1).	96,529	13.0	1.91	83,967	12.2	1.78	180,496	1.85
Totals	542,945	12.4	2.22	215,830	11.3	1.85	758,775	2.12
Total ore produced to October 1, 1955							828,496	2.09

VENTILATION

During the shaft-sinking and early development period, the working places were ventilated by a St. Joe flotation blower which supplied fresh air through a 12-inch Ventube. This blower has a 20-inch impeller and is driven by an 18-horsepower electric motor. Air capacity ranged from a maximum of 4,500 cubic feet per minute to 3,000 cubic feet per minute against a 17-inch static water pressure. The blower was moved later to a drill hole that was cut by one of the development drifts.

As soon as a connection had been made underground with the 24-inch ventilation and escape shaft (figs. 6 and 8), a blower fan powered by a 50-horsepower motor was set up over the hole. This fan, which delivers 14,000 cubic feet per minute of fresh air to the mine against a 14-inch static water pressure, still supplies the major part of the mine ventilation. The direction of airflow underground is controlled to some extent by two portable blowers.

A small outlying stope is ventilated by a 3-horsepower blower that delivers fresh air from the surface through a 10-inch drill hole at a rate of 500 cubic feet per minute.

The haulageway that is being driven northwest from the mine is ventilated in two steps. A 15-horsepower blower supplies fresh air to the haulageway through a 10-inch drill hole at a rate of 1,500 cubic feet per minute. A blower of equal size, mounted in the haulageway, relays the air to the face through a 12-inch Ventube.

Ventilation throughout the mine is sufficient to assure that neither the free silica nor the use of diesel engines underground will be injurious to the health of the men. The dust counts are kept well below the permissible limit of 10 million parts per cubic foot of air by (1) drilling wet at all times, (2) doing no blasting during shifts, (3) using water sprayers attached to the loaders to wet the broken ore as it is being loaded, (4) wetting the ore by sprayers as it is dumped onto the drag conveyer and into the shaft pocket, and (5) using a company-built dust collector at the discharge end of the crusher.

The dust collector is equipped with a 20-horsepower exhaust fan, which has a capacity of 5,000 cubic feet per minute. Intake and discharge lines are 14 inches in diameter. The discharge end of the crusher is enclosed completely to keep dust leakage at a minimum. The dust-laden air from the crusher is water-wetted in the dust collector, then drawn through a series of baffles where the wetted dust settles out. The cleaned air passes out of the collector and is discharged up the shaft. The efficiency of the machine in dust removal is over 70 percent by volume and over 90 percent by weight.

Air contamination from the diesel engines is minimized by the use of high-grade fuel and by water scrubbers on the engine exhausts. The volume and circulation of fresh air in the mine are sufficient to remove quickly any harmful gases that are not eliminated by the scrubbers.

MINE DRAINAGE

Before shaft sinking was begun, a 10-inch hole was churn-drilled close to one side of the proposed shaft. An 8-inch deep-well pump, driven by a 125-horsepower electric motor, was installed in the hole to unwater the ground and drain the shaft as sinking progressed. Water removal by this method (up to 350 gallons per minute) was helpful but inadequate, and it was necessary to use a sinking pump throughout the shaft-sinking operation.

Upon completion of shaft sinking, a connection was made with the drill hole near the bottom of the shaft to permit direct access of the shaft water to the deep-well pump. Additional temporary pumping capacity was provided by two 150-horsepower, electrically driven, multistage centrifugal pumps that were installed over a small sump at the bottom of the shaft. These pumps, each capable of pumping water to the surface at a rate of 330 gallons per minute, were operated during the working shifts, and the deep-well pump was used between shifts. The inflow of water to the shaft, mainly from the sandstone that underlies the ore body, was 375 gallons per minute immediately after the shaft was bottomed and increased gradually as development progressed.

The lowest part of the ore body, southeast of the shaft, was developed first to provide drainage for the mine. A permanent pump station and sump were excavated there, and two 10-inch holes were churn-drilled to the sump area. Two 3-stage centrifugal pumps were installed in the pump station, and an 8-inch deep-well pump was set in one of the drill holes for use as an auxiliary or emergency pump. The other drill hole is used as an outlet from the other pumps (fig. 8). One of the centrifugal pumps is driven by a 200-horsepower electric motor and has a capacity of 900 gallons per minute, whereas the other is driven by a 150-horsepower motor and has a capacity of 550 gallons per minute. The inflow of water to the sump is approximately 1,150 gallons per minute, and the pumping lift is 600 feet. The sump has a capacity adequate for 48-hours storage in the event of a forced shutdown of the mine pumps.

A 4-inch centrifugal pump is used to remove seepage water from the bottom of the shaft. Water is pumped to the mine level and flows from there to the main sump.

WAGE AND BONUS SYSTEM

Each employee is paid a guaranteed base wage for each shift worked plus a daily lead bonus, which is determined by the market price of lead. In addition, an incentive bonus system has been set up whereby employees who are performing certain kinds of work can increase their earnings by added effort and efficiency.

The daily lead bonus in 1953 averaged $1.0625. The wage scales in the Lead Belt during the same period were as follows:

Occupation	Guaranteed daily wage
Drillers (development and mining)1/	$12.18
Loader operators1/	13.18
Scraper operators1/	12.18
Shovel operators (development)1/	12.18
Shuttle-car operators1/	12.12
Motormen1/	12.12
Truck driver (underground)1/	12.12
Skip loader	12.18
Hoist engineer	12.18
Pumpman	12.12
Roofman	12.18
Handyman	12.18
Jackhammer prospector	12.18
Maintenance men	12.35-13.18
Laborer (underground)	12.12
Laborer (surface)	10.35
Shaftman	15.15

1/ Employees in these occupations also participate in the incentive bonus plans.

Development bonuses are paid for measured or equivalent footage of advance at various prices per foot. Unit prices are predetermined for all types of drifts, inclines, raises, side slabbing, back slabbing, and ditching. Stope bonuses are paid at various rates per ton of ore mined. Rates are established for different heights of breast, back and bottom stoping, and for scraping and chute pulling. Parts of the costs of drilling labor and blasting materials are charged to the development and stope projects, but no charge is made for other supplies. Allowance was made for normal operating delays when the unit prices were set. Time spent on moving to a new location and in preparatory work is not charged to the project.

Loading bonuses are paid for loading and transporting ore from stopes to the underground crusher. Bonus payments are made for tonnages of ore moved per shift (weekly average) in excess of set standards. A loading crew consists of a loader operator and a truck or shuttle-car driver. Crew members are responsible for preventive maintenance and help with minor repairs.

A separate bonus is computed for each development, stope, and loading crew. Bonuses, which are paid weekly, are shared by the crew members. The incentive bonus system has been used in the Lead Belt for many years and has proved to be effective both in stimulating increased unit output and increasing employee earnings.

SAFETY AND HYGIENE

The St. Joseph Lead Co. has a well-organized Safety and Hygiene Department. Safety is primarily the responsibility of the foreman, but the Safety Department staff is always available to give needed assistance and is responsible for execution of the company safety and hygiene program throughout the Lead Belt.

One fatal accident occurred during the sinking of the Hayden Creek shaft, but since then (1950) the safety record has been good. During the years 1951-55, the frequency rating (lost-time accidents per million man-hours worked) at Hayden Creek has averaged 36.0, and the severity rating (shifts lost per 1,000 man-hours worked) averaged 2.0. These figures are slightly above the average for the Lead Belt but compare favorably with the national averages for all mines. Accident data for the Hayden Creek mine from June 1, 1949, to June 30, 1955, are shown in table 4.

TABLE 4. - Breakdown of accidents by causes and years, in terms of number of lost-time (L.T.) and non-lost-time (N.L.T.) accidents

Shaft Sinking

Causes	1949		1950		1951	
	June 1 - Dec. 31		Year		Jan. 1 - June 30	
	No. of L.T.1/	No. of N.L.T.	No. of L.T.1/	No. of N.L.T.	No. of L.T.	No. of N.L.T.
Hand tools	0	3	0	2	0	1
Falling objects	1 (14)	0	1 fatal	1	0	0
Cage, skip, bucket	0	2	2 (200)	0	0	1
Other	0	13	1 (8)	14	0	10
Total	1 (14)	18	1 fatal 3 (208)	17	0	12

Development and Mining

Causes	1951		1952		1953		1954		1955	
	July 1 - Dec. 31		Year		Year		Year		Jan. 1 - June 30	
	No. of L.T.	No. of N.L.T.	No. of L.T.1/	No. of N.L.T.	No. of L.T.1/	No. of N.L.T.	No. of L.T.1/	No. of N.L.T.	No. of L.T.1/	No. of N.L.T.
Falling rock	0	0	0	0	1 (202)	1	0	3	1 (45)	4
Loading rock	0	2	0	0	0	0	0	1	0	1
Hand tools	0	1	0	1	0	8	1 (9)	8	1 (138)	5
Explosives	0	0	0	0	0	0	0	0	0	0
Hauling	0	0	0	0	0	3	0	0	0	1
Slips, falls	0	0	0	1	1 (144)	1	0	4	1 (4)	0
Drilling	0	1	0	7	0	15	2 (29)	17	0	6
Electricity	0	0	0	0	0	0	0	1	0	0
Machinery	0	0	0	0	1 (7)	1	0	1	0	0
Material handling ...	0	1	1 (45)	1	0	3	0	3	0	1
Other	0	4	0	8	0	5	0	7	2 (76)	3
Total	0	9	1 (45)	18	3 (353)	37	3 (38)	45	5 (263)	21

1/ Figures in parentheses denote number of shifts lost.

TABLE 5. - Development crew and equipment

Period worked: Aug.-Dec. 1952 Average advance: 22 feet per day

Job classification	No. men	Operating equipment	No. units	Spare equipment	No. units
Driller	8	2-5/8-inch rock drills - mounted on 42-inch air feed	4	2-5/8-inch rock drills (42-inch air feed)	2
Hoistman	2				
Skip loader	1			Joy loader	1
Maintenance man	2			12-1/2-ton electric shuttle car	1
Laborer	1	Joy loaders	2		
Trainee shift foreman	1	12-1/2-ton electric shuttle cars.	2		
Mine captain	1				
Total	16				

TABLE 6. - Mining crew and equipment

Period worked: July-Dec. 1953 Average daily ore production: 1,303 tons

Job classification	No. men	Operating equipment	No. units	Spare equipment	No. units
Jumbo driller	17	Joy loader	3	Joy loader	1
Loader operator	6	Diesel truck	3	12-1/2-ton shuttle car.	1
Driver	6	12-1/2-ton electric shuttle car (part time)	1		
Maintenanceman	6				
Crusherman	2				
Hoistman	3	Mobile drilljumbo	5		
Skip loader	2				
Roofman	4				
Stockroom man	1				
Pumpman	3				
Shift foreman	2				
Mine captain	1				
Total	53				

TABLE 7. - Summary of costs in units of labor, power, and supplies

Shaft Sinking

Period worked: May 1949 - July 1951 Depth of shaft: 701 feet

A. Labor (man-hours per foot)		
Breaking (drilling and blasting)		13.30
Mucking (hand and machine)		21.62
Hoisting		9.95
Pumping		8.61
Shaft scaling		3.11
Cutting timber hitches		20.60
Timbering		14.62
General		11.74
Total labor		103.55
Labor, percent of total cost		66.4
B. Power and supplies		
Explosives (60-percent strength) pounds per foot		46.9
Timber cost per foot		$15.61
Total power kw.-hr. per foot		1,290.0
(1) Air compression		
Drilling	223	
Machine mucking	223	
(2) Hoisting	228	
(3) Pumping	358	
(4) Ventilation	143	
(5) Lighting	115	
Other supplies, percent of total power and supplies		53.6
Power and supplies, percent of total cost		33.6

TABLE 8. - Summary of costs in units of labor, power, and supplies

Development

Period worked: Aug.-Dec. 1952

Total advance (equivalent drift footage): 2,334

A. Labor (man-hours per foot)			
Breaking (drilling and blasting)			1.82
Loading rock			.54
Haulage and hoisting			1.64
Supervision			.73
General			1.09
Total labor			5.82
Labor, percent of total cost			31.2
B. Power and Supplies			
Explosives (45-percent semigelatin)		pounds per foot	10.8
Total power		kw.-hr. per foot	618.0
(1) Air compression	119		
(2) Loading	5		
(3) Haulage	17		
(4) Hoisting	44		
(5) Pumping	330		
(6) Ventilation	25		
(7) Lighting	17		
(8) Miscellaneous	61		
Other supplies, percent of total power and supplies			65.8
Power and supplies, percent of total cost			68.8

TABLE 9. - Summary of costs in units of labor, power, and supplies

Mining

Period covered: July - Dec. 1953
Mining method: Open stopes with pillar support
Tons of ore mined: 168,159

A. Labor (man-hours per ton)		
Breaking (drilling and blasting)		0.104
Loading		.037
Haulage and hoisting		.068
Supervision		.018
General		.098
Total labor		.325
Average tons per man-shift (underground)		26.1
Average tons per man-shift (surface)		434.5
Average tons per man-shift (total)		24.6
Labor, percent of total cost		36.2
B. Power and Supplies		
Explosives (45-percent semigelatin) pounds per ton		.87
Total power kw.-hr. per ton		15.10
(1) Air compression	4.28	
(2) Loading and hauling	1.19	
(3) Hoisting	1.19	
(4) Pumping	6.52	
(5) Ventilation	.66	
(6) Lighting	.21	
(7) Miscellaneous	1.05	
Other supplies, percent of total power and supplies		72.5
Power and supplies, percent of total cost		63.8

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